UPGRADING TITANIFEROUS MATERIALS

This invention relates to the removal of impurities from naturally occurring and synthetic titaniferous materials. The invention is particularly suited to the enhancement of titaniferous materials used in the production of titanium metal and titanium dioxide pigments by means of industrial chlorination systems.

Embodiments of the present invention have the common feature of roasting of titaniferous materials in the presence of additives and at temperatures which encourage the formation of a liquid oxide or glassy phase, followed at some stage by cooling and aqueous leaching as steps in an integrated process. Additional steps may be employed as will be described below.

In industrial chlorination processes titanium dioxide bearing feedstocks are fed with coke to chlorinators of various designs (fluidised bed, shaft, molten salt), operated to a maximum temperature in the range 700 - 1200C. The most common type of industrial chlorinator is of the fluidised bed design. Gaseous chlorine is passed through the titania and carbon bearing charge, converting titanium dioxide to titanium tetrachloride gas, which is then removed in the exit gas stream and condensed to liquid titanium tetrachloride for further purification and processing.

The chlorination process as conducted in industrial chlorinators is well suited to the conversion of pure titanium dioxide feedstocks to titanium tetrachloride.

5   However, most other inputs (i.e. impurities in feedstocks) cause difficulties which greatly complicate either the chlorination process itself or the subsequent stages of condensation and purification and disposal of waste. The attached table provides an indication of the types of

10 problems encountered. In addition, each unit of inputs which does not enter products contributes substantially to the generation of wastes for treatment and disposal. Some inputs (e.g. particular metals, radioactives) result in waste classifications which may require specialist disposal in

15 monitored repositories.

Preferred inputs to chlorination are therefore high grade materials, with the mineral rutile (at 95-96% TiO_2) the most

20 suitable of present feeds. Shortages of rutile have led to the development of other feedstocks formed by upgrading naturally occurring ilmenite (at 40-60% TiO_2), such as titaniferous slag (approximately 86% TiO_2) and synthetic rutile (variously 92-95% TiO_2). These upgrading processes

25 have had iron removal as a primary focus, but have extended to removal of magnesium, manganese and alkali earth impurities, as well as some aluminium.

	Elemental Input	Chlorination	Condensation	Purification
5	Fe, Mn	Consumes chlorine, coke, increases gas volumes	Solid/liquid chlorides foul ductwork, make sludges	
10	Alkali & alkali earth metals	Defluidise fluid beds due to liquid chlorides, consume chlorine, coke		
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20	Al	Consumes chlorine, coke	Causes corrosion	Causes corrosion, makes sludges
25	Si	Accumulates in chlorinator, reducing campaign life. Consumes coke, chlorine	Can encourage duct blockage. Condenses in part with titanium tetrachloride	May require distillation from product
30				
35	V			Must be removed, by chemical treatment and distillation
40	Th, Ra	Accumulates in chlorinator brickwork, radioactive; causes disposal difficulties		
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In the prior art synthetic rutile has been formed from titaniferous minerals, e.g. ilmenite, via various techniques. According to the most commonly applied technique, as variously operated in Western Australia, the titaniferous

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mineral is reduced with coal or char in a rotary kiln, at
temperatures in excess of 1100 C. In this process the iron
content of the mineral is substantially metallised. Sulphur
additions are also made to convert manganese impurities
5   partially to sulphides. Following reduction the metallised
product is cooled, separated from associated char, and then
subjected to aqueous aeration for removal of virtually all
contained metallic iron as a separable fine iron oxide. The
titaniferous product of separation is treated with 2-5%
10   aqueous sulphuric acid for dissolution of manganese and some
residual iron. There is no substantial chemical removal of
alkali metals or alkaline earths, aluminium, silicon,
vanadium or radionuclides in this process as disclosed or
operated. Further, iron and manganese removal is incomplete.
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Recent disclosures have provided a process which operates
reduction at lower temperatures and provides for hydrochloric
acid leaching after the aqueous aeration and iron oxide
separation steps. According to these disclosures the process
20   is effective in removing iron, manganese, alkali and alkaline
earth impurities, a substantial proportion of aluminium
inputs and some vanadium as well as thorium. The process may
be operated as a retrofit on existing kiln based
installations. However, the process is ineffective in full
25   vanadium removal and has little chemical impact on silicon.

In another prior art invention relatively high degrees of
removal of magnesium, manganese, iron and aluminium have been
achieved. In one such process ilmenite is first thermally
30   reduced to substantially complete reduction of its ferric
oxide content (i.e. without substantial metallisation),
normally in a rotary kiln. The cooled, reduced product is
then leached under 35 psi pressure at 140-150 C with excess
20% hydrochloric acid for removal of iron, magnesium,

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aluminium and manganese. The leach liquors are spray roasted
for regeneration of hydrogen chloride, which is recirculated
to the leaching step.

5   In other processes the ilmenite undergoes grain refinement
by thermal oxidation followed by thermal reduction (either
in a fluidised bed or a rotary kiln). The cooled, reduced
product is then subjected to atmospheric leaching with excess
20% hydrochloric acid, for removal of the deleterious
10   impurities. Acid regeneration is also performed by spray
roasting in this process.

In all of the above mentioned hydrochloric acid leaching
based processes impurity removal is similar. Vanadium,
15   aluminium and silicon removal is not fully effective.

In yet another process ilmenite is thermally reduced (without
metallisation) with carbon in a rotary kiln, followed by
cooling in a non-oxidising atmosphere. The cooled, reduced
20   product is leached under 20 - 30 psi gauge pressure at 130°C
with 10 - 60% (typically 18 - 25%) sulphuric acid, in the
presence of a seed material which assists hydrolysis of
dissolved titania, and consequently assists leaching of
impurities. Hydrochloric acid usage in place of sulphuric
25   acid has been claimed for this process. Under such
circumstances similar impurity removal to that achieved with
other hydrochloric acid based systems is to be expected.
Where sulphuric acid is used radioactivity removal will not
be complete.
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A commonly adopted method for upgrading of ilmenite to higher
grade products is to smelt ilmenite at temperatures in excess
of 1500°C with coke addition in an electric furnace,
producing a molten titaniferous slag (for casting and

crushing) and a pig iron product. Of the problem impurities only iron is removed in this manner, and then only incompletely as a result of compositional limitations of the process.

In another process titaniferous ore is roasted with alkali metal compounds, followed by leaching with a strong acid other than sulphuric acid (Australian Patent No. AU-B-70976/87). According to this disclosure substantial removal of various impurities is achieved, with "substantial" defined to mean greater than 10%. In the context of the present invention such poor removal of impurities, especially of thorium and uranium, would not represent an effective process. No specific phase structure after roasting is indicated for this process but it is evident from analytical results provided (where product analyses, unlike feed analyses do not sum to 100% and analyses for the alkali metal added are not given) that there may have been significant retention of the additive in the final product. Under the conditions given it is herein disclosed that it is to be expected that alkali ferric titanate compounds which are not amenable to subsequent acid leaching will form. The consequent retention of alkali will render the final product unsuitable as a feedstock for the chloride pigment process.

In yet another process a titaniferous ore is treated by alternate leaching with an aqueous solution of alkali metal compound and an aqueous solution of a mineral acid (US Patent No. 5,085,837). The process is specifically limited to ores and concentrates and does not contemplate prior processing aimed at artificially altering phase structures. Consequently the process requires the application of excessive reagent and harsh processing conditions to be even partially effective and is unlikely to be economically

implemented to produce a feedstock for the chloride pigment
process.

A wide range of potential feedstocks is available for
upgrading to high titania content materials suited to
chlorination. Examples of primary titania sources which
cannot be satisfactorily upgraded by prior art processes for
the purposes of production of a material suited to
chlorination include hard rock (non detrital) ilmenites,
siliceous leucoxenes, many primary (unweathered) ilmenites
and large anatase resources. Many such secondary sources
(e.g. titania bearing slags) also exist.

Clearly there is a considerable incentive to discover methods
for upgrading of titaniferous materials which can
economically produce high grade products almost
irrespectively of the nature of the impurities in the feed.

The present invention provides a combination of processing
steps which may be incorporated into more general processes
for the upgrading of titaniferous materials, rendering such
processes applicable to the treatment of a wider range of
feeds and producing higher quality products than would
otherwise be achievable.

Accordingly, the present invention provides a process for
upgrading a titaniferous material by removal of impurities
which process includes the steps of:-

    (i)   heating a titaniferous material to a
temperature less than 1300°C to produce a solid
titaniferous phase and a liquid oxide or glassy
phase in the presence of sufficient of
compounds which encourage the formation of the
liquid oxide or glassy phase;

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(ii)   cooling the product of step (i) to form a
        solidified material comprising the titaniferous
        phase and an impurity bearing phase at a rate
5   sufficient to ensure the susceptibility of the
        impurity bearing phase to leaching in either an
        acid or alkaline leachant; and

(iii)   leaching the solidified material in an acidic
10   or alkaline leachant to leach at least a
        portion of the impurities.

In order to ensure the formation of the solid titaniferous
phase and the liquid oxide or glassy phase during the heating
15 step it will normally be necessary to add to the titaniferous
material, prior to the heating step, sufficient of a compound
that encourages the formation of the liquid oxide or glassy
phase. However, in some cases it will not be necessary since
the titaniferous material itself may contain sufficient of
20 such a compound.

It has been discovered that the process of the invention can
remove iron, magnesium and other alkaline earths, alkalis,
manganese,   silica, phosphorus, alumina, vanadium, rare
25 earths, thorium and other radioactive elements, which
impurities form an almost comprehensive list of impurities
in titaniferous mineral sources.   From most materials a
product purity of greater than 96% TiO_2 can be obtained.

30 Compounds added to the titaniferous material may be mixed
therewith by any means ranging from direct mixing of
additives prior to charging to thermal treatment to more
complex feed preparations such as the formation of
agglomerates or nodules of mixed products, to briquette

production from feeds and additives. Many additives will be effective. In particular it is herein disclosed that sodium, potassium, lithium, phosphorus, silicon and boron compounds and minerals (e.g. borax, trona and other alkali metal

5   carbonates, spodumene, caustic soda) will be effective. Additives may be incorporated individually or in combination with other additives.

It is further disclosed herein that the formation of a glassy

10   phase by addition of alkali compounds can be achieved without the formation of alkali titanate phases, reduced alkali titanate phases (e.g. $NaTiO_2$ - compounds and solid solutions) or alkali ferric titanate phases (e.g. $Na(Fe, Al)O_2$ - TiO_2 phases known as "bronzes") in roasting. Where such titanate

15   phases form their stability with respect to subsequent leaching steps is such that the final product quality is adversely affected. The incorporation of sufficient quantities of further additives (e.g. boron or phosphorus compounds) which substantially reduce alkali oxide chemical

20   activity can have the effect of eliminating these phases.

Under many circumstances it will be beneficial to incorporate multiple additives into the material to be treated by thermal processing. For example, it is herein disclosed that the

25   simultaneous presence of silica, anhydrous borax and sodium oxide in 1000°C thermally processed material in weight ratios of about 7:1:1 ensures the preferential formation of a glassy phase over other phases containing silica or soda. In this formulation the required borax addition is only just over 10%

30   of the addition which would be required for an equivalent amount of glassy phase where other additives do not act as extenders. Since borax is by far the most expensive additive of the three additives in this case optimum economics are achieved by the use of the extenders.

Thermal processing may be carried out in any suitable device. The production of liquid phases would recommend rotary or grate kilning, but shaft furnaces may also be used and it has

5   been found that fluidised beds can be used under some circumstances. Any gaseous atmosphere conditions may be used, from fully oxidising to strongly reducing. The thermal processing atmosphere should be chosen to most suit other steps in integrated processing. Reducing conditions may be

10   achieved where desired by either the use of a sub stoichiometric firing flame or the addition of coal, char or coke with the thermal processing charge.

Thermal processing residence time at temperature will depend

15   on the nature of the additives and the feed, as well as the operating temperature. Residence times of from 5 minutes to five hours have been effective, allowing thermal processing residence times to be set to most suit other requirements in integrated processing.

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The level of additive used and the conditions applied in thermal processing should be such that glassy phase formation does not exceed the limitations set by materials handling constraints in the thermal processing step. For example,

25   where glassy phase formation exceeds about 15% by volume of the roasted material it must be anticipated that accretion and bed fusion problems will occur.

Cooling of the thermally treated material should be conducted

30   in such a manner as to limit the reversion of the glassy phase to crystalline phases, i.e. should be at a sufficient rate to a temperature at which the liquid glass solidifies as to ensure the formation of at least a portion of solid glass rather than complete formation of crystalline products.

Further, cooling should be conducted under an environment appropriate to the conditions of thermal treatment (i.e. reduction processing will require cooling in an oxygen free environment).

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The aqueous leaching step need not necessarily follow directly after the presently disclosed thermal processing step. For example if the thermal processing step is conducted under oxidising conditions it may be optionally followed by 10 a reduction step prior to aqueous leaching. Further, crushing/grinding of the thermally processed material to enhance subsequent leach performance may be undertaken.

The conditions necessary to conduct effective leaching will 15 depend on the nature of the original feed and the additives. For example, addition of soda ash and borax to siliceous leucoxene in accordance with the present disclosure will result in a product which can be leached in sodium silicate solution formed directly from the thermally treated material; 20 the active leachant in this case is simply water. In other cases up to 100 gpL caustic soda solution or acid will be an effective leachant. Leaching will generally benefit substantially by use of high temperature (e.g. 80°C or above), although it has not been necessary to use pressure 25 leaching to achieve effective conditions. Nevertheless it is presently disclosed that pressure leaching can be effectively and successfully applied. Lower temperatures can also be used, although with penalties in process kinetics.

30 Leaching may be conducted in any circuit configuration, including batch single or multiple stage leaching, continuous cocurrent multistage leaching, or continuous countercurrent multistage leaching. For most circumstances two stage cocurrent leaching will be most beneficial. Average residence

time may vary from 30 minutes to 10 hours, depending on process conditions. Any leach vessel capable of providing adequate shear may be used. Simple stirred tank vessels are applicable.

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At the conclusion of leaching the leach liquor may be separated from the mineral by any suitable means, including thickening, filtration and washing. The mineral product may then pass on to other steps in an integrated process. For

10   example, a further acid leach may follow the disclosed leaching step, particularly where the titaniferous feed has a content of alkalis or alkaline earths.

Other processing steps may be added as necessary or desired.

15   For example, reagent regeneration (e.g. caustic regeneration, hydrochloric acid regeneration, sulphuric acid regeneration) can be used with the process in order to improve process effectiveness or economics. Similarly, a physical separation step may be employed at any stage (e.g. a final magnetic

20   separation to remove grains containing iron, such as chromite).

Examples
Example 1

25   Sodium carbonate addition, corresponding to 4.25% Na_2O by weight, was made to a titania concentrate whose composition is given in Table 1. The mixture was homogenised and pelletised, and the pellets were heated in air to 1000°C for 4 hours. The thus roasted pellets were quenched in liquid

30   nitrogen and then crushed to pass a screen of 200 microns aperture. The crushed roasted pellets were subjected to leaching under reflux with 40 wt% sodium silicate solution ($SiO_2:Na_2O=2,4:1$ by weight) at 4% slurry density. (Sodium silicate solution was used to simulate leaching using water

as leachant under conditions where the leach liquors are recycled to leaching after solid/liquid separation).

5   Solid/liquid separation was effected by centrifuging, after which the leach residue was washed and calcined at 1000°C for analysis. The analysis of the calcined product is also given in Table 1.

The original concentrate was known to contain silica
10   primarily as quartz inclusions in titanate grains. X-ray diffraction analysis after roasting indicated extinction of all crystalline phases containing silica. A glassy phase containing 16% Na_2O, 46% SiO_2, 9% Al_2O_3, 26% TiO_2 and 3% Fe_2O_3 was identified in the roasted material by electron
15   microscopy. Sodium titanates and sodium iron titanium bronze were also identified (along with rutile) by these techniques, indicating that conditions were not optimised.

Nevertheless, highly effective concentrate upgrading has been
20   achieved even where the benefits of subsequent acid leaching have not been pursued, ·illustrating the benefits of the formation of the glassy phase. Substantial removal of silica, alumina and vanadium was achieved.

25   Example 2
This example illustrates the optimisation of additives for both process effectiveness and most economic formulation.

In this example titania concentrates of the composition given
30   in Table 2 were used as titaniferous material for treatment. Early work attempting to produce glassy phase with this material by addition of sodium carbonate prior to roasting indicated that glassy phase could easily be produced, but over a wide range of conditions reduced sodium titanate or

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sodium iron titanate bronze formation which resulted in
sodium retention after leaching could not be easily avoided.

Complete and partial replacement of sodium carbonate by borax
5   was tested.

Two batches of hand pressed pellets were prepared as follows.
A 100g sample of the concentrates (previously ground to
passing a screen aperture of 30 microns) was blended in each
10   case with 1.1% of the appropriate additive or additive
mixture and the resulting blends were pressed into pellets.
The first batch was prepared with 1.1 wt% of anhydrous borax
addition while the second batch was prepared with addition of
1.1 wt% of 1 : 1 $Na_2B_4O_7:Na_2O$.
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Each batch of pellets was roasted for two hours in a 7 : 1
H_2/CO_2 atmosphere at 1000°C and then removed to cool quickly
in the same atmosphere. The roasted pellets were ground to
pass a screen aperture of 75 microns for subsequent leaching.
20   Ground roasted pellets were caustic leached under reflux
conditions for 6 hours in a 10% NaOH solution at 6.7% solids
density. Solid/liquid separation was effected by filtration,
and the caustic leached products were washed and dried in
preparation for subsequent acid leaching.
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The caustic leached residues were acid leached in 15% HCl for
4 hours under reflux, then similarly filtered, washed and
dried.

30   In each case samples of the concentrate and roasted material
were submitted for X-ray diffraction analysis. While quartz
and various ilmenite, anatase and rutile related phases were
identified in the concentrates the only crystalline phases
identified in the roasted product were rutile and ilmenite.

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All quartz had entered a glassy phase, and no titanate phases
which would reduce leach effectiveness were identified.

Analyses of the caustic and acid leach residues in each case,
5   illustrating the effectiveness of the process where optimum
conditions are applied, are provided in Table 3.

Example 3
The same pellet formulations as indicated in Example 2 were
10   made up in 350 kg batches in an agglomeration plant and
roasted at 30 kg/hr feed rate with 15% brown coal char
addition to a final temperature of 1000°C in a small (0.5m
diameter) rotary kiln.   Residence time above 900°C was
approximately 10 minutes.   There were no problems with
15   accretions or bed fusion, and after separation from residual
char the products had exactly the same properties as the
roasted products of Example 2.

Example 4
20   A commercial titania slag product having the composition
indicated in Table 4 was processed as for the processing
conditions indicated in Example 2, but with 2 wt% anhydrous
borax addition in place of the other additives. The caustic
leach was conducted at 165°C under pressure, and a pressure
25   leach with 20% sulphuric acid conducted at 135°C was used in
place of the hydrochloric acid leach. The final residue was
calcined at 900°C for one hour.   The products of this
treatment are indicated in Table 4.

30   **Example 5**
This example when compared with examples 1 and 2 illustrates
the advantages of the formation of a glassy phase.

Concentrates having the composition indicated in Table 1 were

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subjected leaching under atmospheric reflux conditions with excess 20% HCl. After separation of the residue from the liquor followed by washing and drying of the residue its composition was as given in Table 5. Clearly there was ineffective removal of virtually all impurities of interest by comparison with the other examples provided herein.

Table 1: Concentrates and Product from Example 1

wt%	Concentrate	Product
TiO_2	85.8	94.9
Fe_2O_3	2.25	1.91
Al_2O_3	1.08	0.63
SiO_2	7.62	0.74
Nb_2O_5	0.30	0.31
V_2O_5	0.235	0.02
Na_2O	0.0	1.10

Table 2: Composition of Concentrates
Used in Examples 2 and 3

	wt%
TiO_2	63.6
Fe_2O_3	28.6
$SiO2$	3.53
Al_2O_3	0.80
MgO	0.87
CaO	0.02
Cr_2O_3	0.55
MnO	1.11
V_2O_5	0.22
ZrO_2	0.26
P_2O_5	0.04
U_3O_8	0.002
ThO_2	0.01

Table 3: Compositions of Leach Products from Example 2

5	wt%	1.1% $Na_2B_4O_7$ addition		1.1% 1:1 $Na_2B_4O_7$:Na_2O addition	
		Caustic Leach Residue	Acid Leach Residue	Caustic Leach Residue	Acid Leach Residue
10	TiO_2	66.9	94.3	67.3	94.9
	Fe_2O_3	27.1	30.2	30.6	2.04
	SiO_2	1.12	0.99	0.55	0.86
	Al_2O_3	0.22	0.17	0.14	0.15
	MgO	0.97	0.08	0.90	0.09
15	CaO	0.05	0.001	0.03	0.001
	Cr_2O_3	0.68	0.69	0.70	0.67
	MnO	1.15	0.06	1.19	0.06
	V_2O_5	0.22	0.15	0.23	0.13
	ZrO_2	0.27	0.37	0.28	0.38
20	Na_2O	0.05	0.02	0.15	0.03
	P_2O_5	0.02	0.02	0.01	0.02
	U_3O_8	0.002	0.002	0.002	0.002
	ThO_2	0.01	0.003	0.01	0.004

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Table 4:   Feed and Product in Example 4

	wt%	Commercial Slag	Roast/Leach Product
5	TiO_2	79.7	97.2
	FeO	9.24	0.85
	SiO_2	3.11	0.09
	Al_2O_3	3.23	0.38
	MgO	4.81	0.43
10	CaO	0.41	0.002
	Cr_2O_3	0.16	0.12
	MnO	0.25	0.02
	V_2O_5	0.57	0.12
	ZrO_2	0.046	0.06
15	P_2O_5	0.002	0.004
	U_3O_8	0.0005	n.d.
	ThO_2	0.0006	n.d.

n.d. = not determined

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Table 5:   Results of Processing as described in Example 5

	wt%	Leach Product
	TiO_2	88.6
25	Fe_2O_3	0.98
	SiO_2	7.54
	Al_2O_3	0.65
	V_2O_5	0.198
	U_3O_8	0.0054
30	ThO_2	0.0094